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BUREAU OF MINES
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INFORMATION CIRCULAR

GOLD MINING AND MILLING METHODS AND COSTS AT THE GOLD HILL MINE OF TALACHE MINES, INC., QUARTZBURG, IDAHO

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BY

JOE H. SKIDMORE

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GOLD MINING AND MILLING METHODS AND COSTS AT THE GOLD HILL MINE OF TALACHE MINES, INC., QUARTZBURG, IDAHO[1]

By Joe H. Skidmore[2]

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2/ One of the consulting engineers, Bureau of Mines, and superintendent, Talache Mines, Inc., Quartzburg, Idaho.

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INTRODUCTION

This paper, describing mining and milling methods and costs at the Gold Hill mine, is one of a series being published by the Bureau of Mines.

Boise Basin, in which the Gold Hill mine is located, is in the central part of Boise County, Idaho, about 30 miles by air line northeast of Boise. The mine, owned and operated by Talache Mines, Inc., is at the southern end of the town of Quartzburg.

About 115 to 125 tons daily is mined by square-setting and is treated in an amalgamation plant.

ACKNOWLEDGMENT

The assistance of A. H. Burroughs, Jr.,3/ in compiling data for this paper is acknowledged.

HISTORY

The Gold Hill vein was discovered in 1863, a year after placer gold was discovered in the Boise Basin. Some of the claims were patented as early as 1870. The Gold Hill vein was mined over a continuous length of 3,000 feet and was productive to a depth of 300 feet. By 1900, the known oreshoots in this vein were so nearly exhausted that the mine was abandoned. The deepest workings at that time were on the 400 level, 404 feet below the collar of the shaft.

The mine was unwatered in 1908 and operated until 1926. The bulk of the production for the period came from the porphyry ore bodies located in the Pioneer and Gold Hill rhyolite dikes. During this period the Pioneer shaft was sunk to the 600 level and the ore stoped to the surface.

After a thorough study of the geology and ore occurrence Talache Mines, Inc., acquired the property in 1927. Although the ore bodies had declined in both size and grade and the bottom level looked poor, it was thought that these apparently discouraging conditions were of a local nature and that good ore similar in character to that produced above the 600 level would be found at greater depths.

3/ President, Talache Mines, Inc.

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Accordingly, the mine was unwatered and the shaft sunk to the 740-foot level. Better-grade ore was encountered and milling operations were begun in 1929 in a newly constructed mill. The tonnage milled was gradually increased from the initial 30 tons per day to 70 tons per day. Operations in this mill continued until August 20, 1931, when the entire surface plant was destroyed by a forest fire. During this period the tonnage milled, which came from the 700 and 900 levels, totaled 43,600 tons averaging $8.03 per ton.

Early in 1934 a 25-ton mill was completed, and a year later its capacity was increased to 100 tons. Recent changes and improvements in the mill have made possible a maximum of 150 tons per day and a continuous average of 115 to 125 tons per day.

At present (1937) the ore bodies in the Pioneer and Gold Hill rhyolite dikes have been mined to the 1100 level.

GEOLOGY

The geology of the district has been described by Ballard[4] and Ross,[5] geologists for the Idaho Bureau of Mines and Geology.

The Gold Hill mine is situated in what is locally known as the porphyry belt. In the mine workings the granite of the Idaho batholith is intruded by several rhyolite, diorite, and small diabase dikes. The two major, ore-bearing rhyolite dikes are the Gold Hill and Pioneer. These irregular dikes range in width from a few feet to a maximum of 150 feet, strike N. 70° to 80° E., and dip steeply to the north. As the Gold Hill rhyolite, or north dike, has a steeper dip than the Pioneer, these two dikes are gradually dipping into each other. Distances between them on the 500, 900, and 1100 levels are 300, 150, and 30 feet, respectively.

The major fissure in the mine workings is the Gold Hill vein, which was formed after the intrusion of the dikes. It strikes N. 70° to 75° E. and dips steeply to the south. The first work in the mine was on this fissure; no ore has been found in the fissure below the 400 level except at intersections with known oreshoots. The fissure shows some heavily mineralized spots on the 1100 level and in places carries values sufficiently high to be considered ore. A drift to the west will explore the ore possibilities of the vein at this horizon.

4/ Ballard, Samuel M., Geology and Gold Resources of the Boise Basin, Boise County, Idaho: Idaho Bur. Mines and Geol. Bull. 9, 1924, pp. 51-58.

5/ Ross, Clyde P., Some Lode Deposits in the Northwestern Part of Boise Basin, Idaho: Geol. Survey Bull. 846d, in cooperation with Idaho Bur. Mines and Geol., 1934, pp. 253-262.

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Several subordinate fissures are known to exist on the property, but their production, confined to shallow surface workings, has been small.

The ore bodies in the two major rhyolite dikes do not seem to bear any direct relation to each other, but they follow fissures in the same regional fracture system. Although no connection between these individual ore bodies and the Gold Hill vein has been traced, their general similarity in strike suggests a mutual relation to the general fracture system of the camp. There seems to be no definite proof that the major fissure was the master channel for solutions to permeate the Pioneer and Gold Hill rhyolite-dike fissures.

The porphyry ore comprises small quartz veinlets in an altered and fractured zone. A quartz seam 1 inch wide is considered as a strong veinlet. A 5-foot face in good ore assaying $10 per ton may contain several knife-blade seams or one or two larger seams ranging in width from 1/4 to 1 inch. Sericite is the typical alteration product of the rocks in the ore zone. Pyrite crystals are sparsely disseminated through the altered rock and in fractures but are not an important locus of gold.

Metallic minerals are disseminated through the quartz in the seams and occasionally in the altered country rock. The metallic minerals identified in the ore include pyrite ($Fe\ S_2$), sphalerite (Zn S), galena (Pb S), galenobismutite ($PbS\ Bi_2\ S_3$), tetradymite ($Bi_2\ Te\ S_2$), tetrahedrite ($Cu_8\ Sb_2\ S_7$), and native gold.

Gold is the principal economic metal recovered and occurs chiefly in the free state. The gold is found most commonly in white quartz veinlets associated with bismuth and iron sulphides. The presence of bismuth usually indicates good gold values. Crushed rhyolite porphyry is the principal gangue material.

No apparent chemical or physical change takes place in the broken ore lying in the stopes. Mill feed carries from 3 to 6 percent moisture.

The ore bodies are irregular-shaped lenses 3 to 40 feet wide, 50 to 300 feet long, and 100 to 800 feet deep, and they occur in a shear zone 200 to 300 feet wide.

MINING

Prospecting and Exploration

Virtually all of the present prospecting is done underground on the three lower levels. The chief mineralization follows a fractured zone 200 to 300 feet wide which includes the two rhyolite dikes. As most of the ore produced is mined from the oreshoots in the dikes, these are followed on the strike and are crosscut at close intervals. The most favorable seams, whether knife-blade or wider, are prospected horizontally and vertically. Because the ore is confined to numerous small quartz seams with no apparent walls, close daily checking and sampling are necessary to determine definitely the pay limits.

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Sampling

As the ore within a shoot varies in grade with the entrance and departure of the small veinlets, the practice is to sample every set of ore in a stope before it is mined. All drifts are sampled across the full face every 5 feet of advance. Crosscuts are sampled on either or both sides for a length of 5 feet per sample. Since the ground is fairly soft, 5- to 10-pound samples are taken with a prospecting pick on a canvas catch- all and transferred to 9- by 14-inch canvas sacks. To make sure every face is sampled the services of the regular sampler were dispensed with, and the sampling job was turned over to the two shift bosses. Each shift boss samples each new face on his shift and sees to it that the samples reach the drier.

Development and Mining

General

The property has been developed and is now being worked through a main vertical shaft sunk to a depth of 1,250 feet. Working levels are established at the 100-, 250-, 400-, 500-, 600-, 700-, 900-, 1,100-, and 1,250-foot horizons. The 1,250 level is just a station at present (Feb. 1, 1937) as the new lift in the shaft has just been completed. The three-compartment shaft is 5 feet 2 inches by 13 feet inside timbers. It is timbered with 10- by 10-inch square timbers and 6- by 10-inch dividers. The manway compartment is 5 feet 2 inches by 5 feet in the clear; each hoisting compartment is 3 feet 6 inches by 5 feet 2 inches in the clear. Hoisting is done in two compartments; the third compartment is used for ladderway, pipe lines, and electric cables.

Shaft Sinking

Shaft sinking is carried on through the manway compartment; a sinking bucket and an automatic dump are used. A 15-horsepower electric hoist, equipped with a 5/8-inch cable and having a rope speed of 300 feet per minute, handles all the spoil, men, and materials. A No. 3 Cameron vertical plunger pump, equipped with 15 feet of 2-inch suction hose and a foot valve covered with common porch screen, handles the water in the bottom. A No. 6 Cameron pump, similar to the no. 3 pump, is always kept in the shaft near the bottom as a stand-by.

When the shaft has been deepened 30 feet below a level, guides are installed in the outside hoisting compartment; these permit shaft timbers to be lowered on a cage without a transfer from the surface to the bottom.

A standard wedge-cut shaft round 5 feet deep, comprising 20 to 24 holes, is blasted with no. 8 instantaneous and delay electric blasting caps and 40-percent, special-gelatin, 1-1/8- by 8-inch powder. No set cycle of operations is maintained. All drilling is done with 55-pound jack-hammers. Each shift relieves the other in the bottom and carries on the work without

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interruption. One hoistman and two miners per shift on the bottom, with an extra helper on day shift for installing guides and pipe and maintaining the pump, comprise the total shaft crew of 10 men per 24-hour cycle. Progress of 2.5 feet per day was maintained for the 172 feet advanced.

The automatic bucket dump used for sinking is shown in figure 1. Equipment is installed in the manway compartment, thus permitting the free and unobstructed use of the two hoisting compartments. Waste from the bottom is hoisted and automatically dumped into cars through a chute extending over the track. The bucket is suspended under a skeleton cross head by two heavy chains on either side. Men and miscellaneous materials are lowered in the bucket. All timbers from the collar are lowered to the bottom in a cage operating through the outside hoisting compartment.

A heavy trap door, made in two sections and free to swing into place on hinges, acts as a bulkhead when the outside compartment is not in use. This trap door, as well as the solid bulkhead in the middle compartment, is placed one set below the level from which sinking operations are started.

Drifting and Crosscutting

Standard drifts and crosscuts are driven 4 by 7-1/2 feet in the clear with 6-inch batter per post. A miner and mucker comprise a crew for this type of work. Mucking and tramming is done by hand. Drilling is done with mounted and unmounted 55-pound jackhammers which use 7/8-inch hollow hexagonal steel and mounted 128-pound drifters which use steel of the same size. Blasting is done with safety fuse, no. 8 blasting caps, and 30-percent special gelatin powder. When speed is essential a continuous three-shift program is adopted. During normal operations work is executed on a one- or two-shift basis.

Raising

All raises are timbered with standard framed square-sets of round timbers. The chute compartment or compartments are 4 feet 8 inches by 5 feet, center line to center line, and the manway is 4 feet 8 inches by 5 feet 10 inches. All sets are 7 feet high center line to center line. The stope-raise set is shown in figure 2. Increasing the length of the manway compartment 10 inches made room for a timber slide 6 inches wide, larger ventilating pipe, and more convenient arrangement of ladder platforms and ladders. Standard raise rounds are drilled with wet stoppers using 1-inch, quarter-octagon, hollow steel. Blasting is done with safety fuse, no. 8 blasting caps, and 30-percent special gelatin powder.

Stoping

A square-set stoping method is used throughout the mine. This method provides maximum support to the back and walls and fills the need for a flexible method of mining. Waste filling is kept as close as possible to the mining breast.

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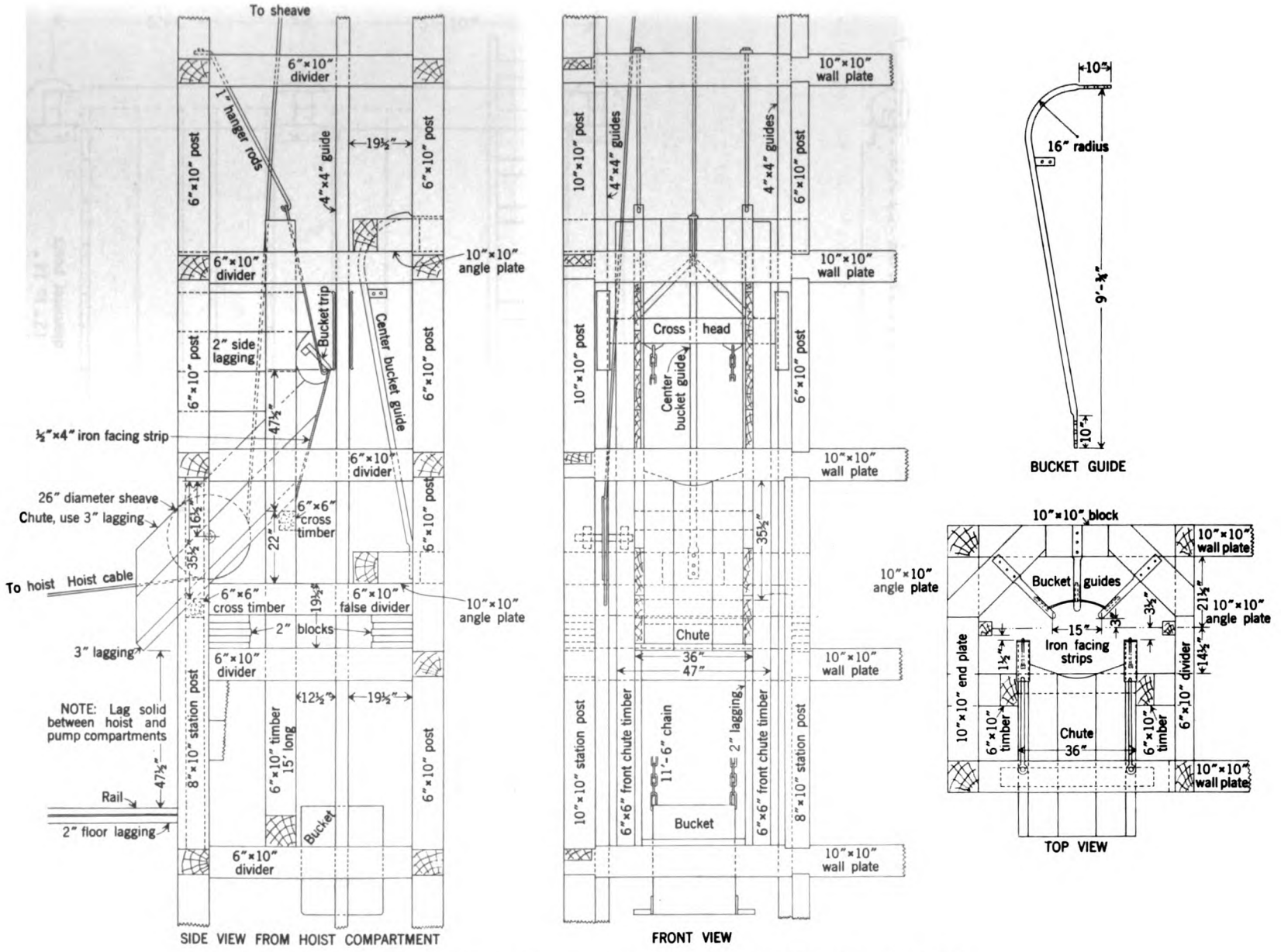

Figure 1—Plan of automatic bucket dump used in sinking from 1100 to 1250 level, Talache Mines, Inc.

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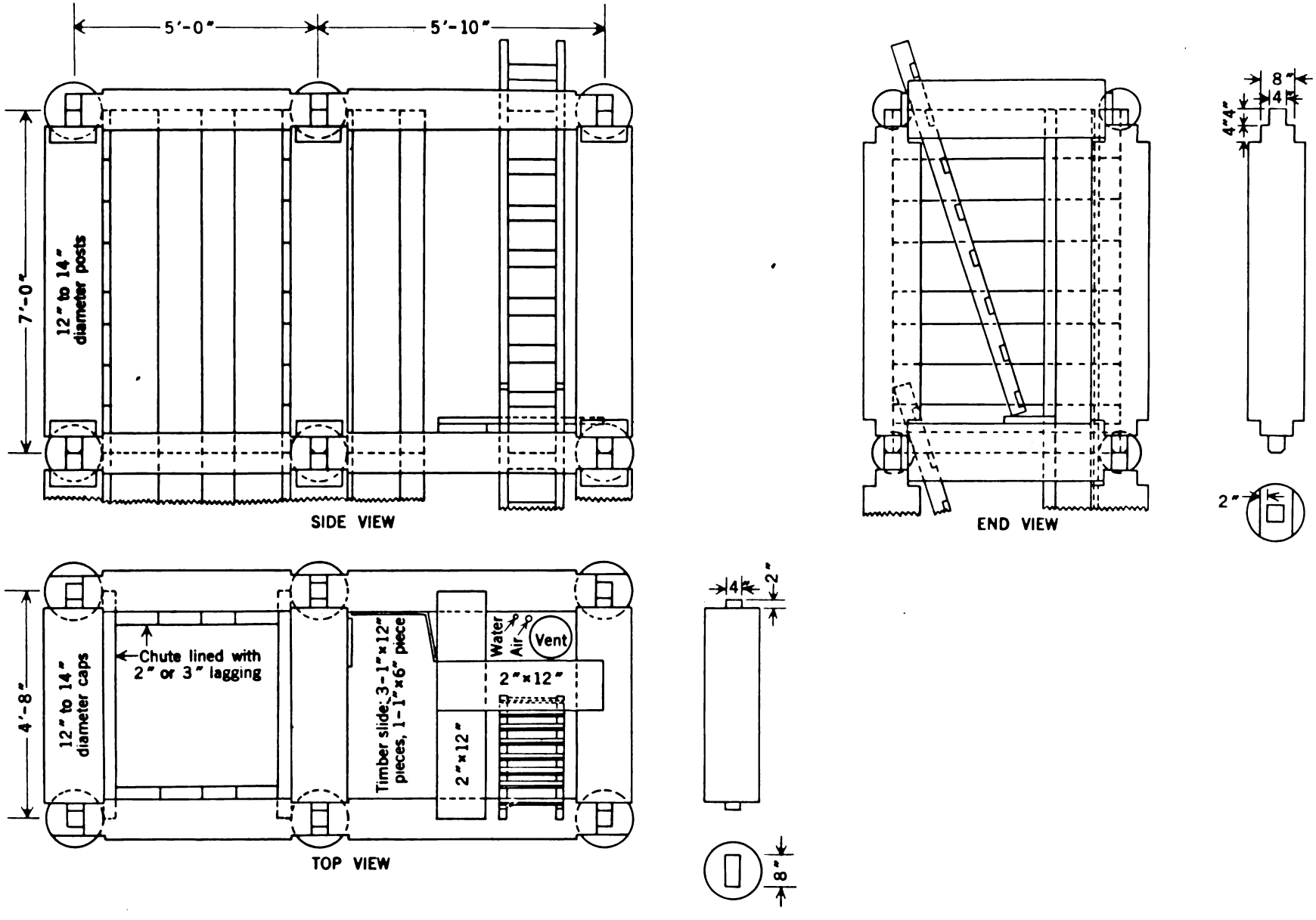

Figure 2.—Stope-raise set, chute and manway construction, Talache Mines, Inc.

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Preparatory to stoping, a drift is driven the entire length of the ore body to be mined. The drift is timbered with round framed timbers having 7-1/2-foot posts. These sets are framed from timbers ranging from 15 to 18 inches in diameter. The sill and second floors are carried one set wide. If the ore is more than one set wide on the third floor the stope is timbered to include all the ore. This system protects and supports the sill floor. The ore so left as longitudinal pillars is stoped after the ore above the third floor has been mined to the level above. Poling to carry the fill is placed on top of the second-floor caps. This leaves one open set above the main drift to facilitate relieving timbers and lagging in heavy ground.

Chutes are placed at 30- to 50-foot intervals, timbered with round framed timbers 12 to 15 inches in diameter, and lined with 2- and 3-inch vertical lagging. A manway is carried up with each chute, thus providing access to the ore pass if repairs are needed or ore hangs up. Figure 2 shows details of stope sets and stope-raise sets. Standard-size timbers used for square-set posts and caps range from 9 to 14 inches in diameter.

All stopes are filled. Sorting within the stopes provides 10 to 20 percent of the fill used. The remainder of the fill usually comes from prospect crosscuts and inclined raises driven from the stopes. Occasionally, it is possible to dump the waste from exploration headings directly into the stopes for fill.

As the average ore is not hard, 90 percent of all drilling is done with hand-held, 55-pound wet jackhammers. Wet stopers are used as the occasion demands. A miner and mucker comprise a crew; each crew is supposed to blast, timber, and muck out one square-set per shift.

Small air hoists, of 5- to 10-horsepower capacity, are used for hoisting materials and supplies into the stopes. Where possible a lead raise is driven in ore to the level above and used to lower timbers into the stope after the lift is approximately one-half worked out.

Underground Transportation

All loading and tramming is done by hand. Ore is transported in flat-bottom, end-dump, 18-inch gage cars fitted with Timken bearings. The cars, which have a capacity of 1 ton each, are hoisted to the surface on cages operating in balance. The ore is dumped directly into the coarse-ore bin of the mill. Twenty-pound rails are used on all important haulageways.

Wages

All work underground is done on days pay. The wage rate per shift for shaft miners is $5.75, for regular miners and pumpmen, $4.50, and for trammers and muckers, $4.00. On the surface the hoistmen are paid $5.00, carpenters $5.25, timber framers $4.50, topmen $4.00, and a blacksmith $5.25.

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Ventilation

Fresh air is forced into the mine from the surface through a 26-inch diameter, 14-gage, electric-arc-welded steel pipe by means of a 15-horsepower, 3,600-r.p.m. blower. Smaller blowers pick the fresh air up from the primary ventilating system on the lower levels and deliver it to faces needing air; 8- and 12-inch, flexible, heavy-rubber-coated tubing is used to transport the air from the smaller fans.

Organization

Figure 3 shows the administrative organization of the company.

Cost of Mining

Table 1 shows the underground costs per dry ton of ore hoisted for September 1936.

Costs are summarized in units of labor, power, and supplies in table 2.

MILLING

The present milling practice comprises straight amalgamation. Prior to 1920 the cyanide process was used to treat the table concentrates. This treatment was very effective when the solutions were fresh and uncontaminated but soon lost its effectiveness as the solution fouled. Because it was found to be more economical to amalgamate with the aid of finer grinding the cyanide process was discarded.

The present mill was built in 1934 as an addition to the 25-ton stamp mill constructed in 1933. Although the original design called for a capacity of 100 tons per day it has been possible to treat as much as 150 tons per 24 hours.

Water for milling is taken from a nearby mountain stream, stored in a reservoir, and conveyed to the mill via open ditch, flume, and iron-pipe penstock. Figure 4 is a flow sheet of the present mill.

Crushing

Ore, of approximately 10 inches maximum diameter, is delivered directly to the coarse-ore bin in 19-cubic foot mine cars hoisted in cages. The receiving bin and upper landing is part of the vertical-shaft head frame.

The ore from the coarse-ore bin is fed by an apron feeder to a 1-inch grizzly. The oversize from the grizzly is crushed in a 10- by 20-inch jaw crusher set to 1-1/2 inches. The fines and crusher product drop by gravity to the hopper of a bucket elevator. The dry elevator is 41 feet high between the center lines of the pulleys. It has 57 buckets, 12 by 7-1/4 inches, spaced 20 inches apart; the bucket speed is 370 feet per minute. This elevator discharges directly into a 145-ton fine-ore bin.

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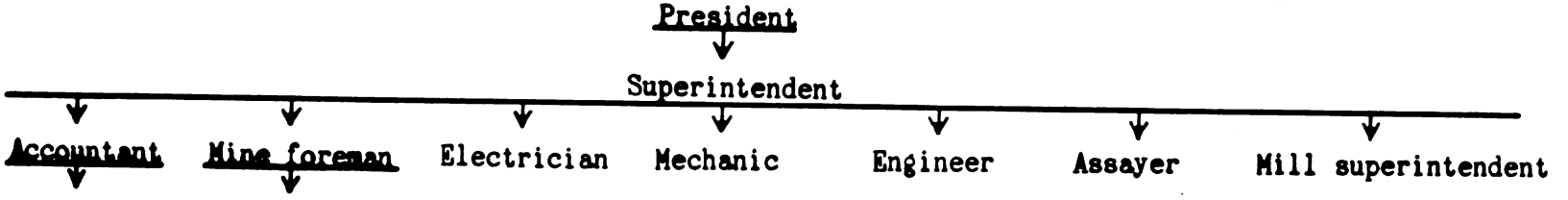

Figure 3.- Administration organization. Talache Mines, Inc.

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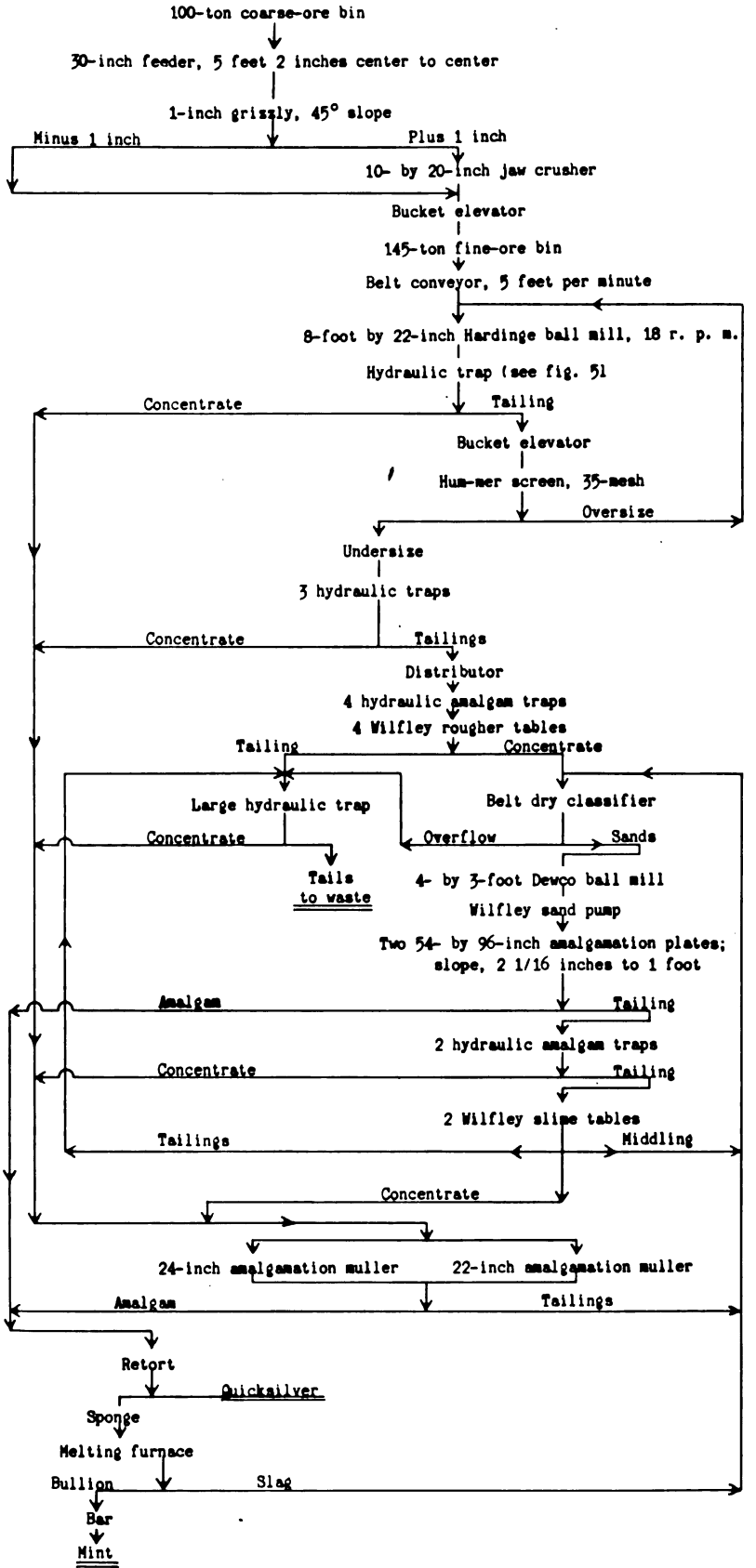

Figure 4.- Mill flow sheet (capacity, 125 tons), Talache Mines, Inc.

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TABLE 1. - Underground costs per dry ton of ore hoisted, September 1936

(Ore hoisted, dry tons, 3,579; ore and waste hoisted, dry tons, 3,919; ore broken, dry tons, 3,700)

	Labor	Super-vision	Compressed air drills and steel	Power	Explo-sives	Timber	Other supplies	Total	Per ton of ore and waste hoisted
Exploration.....	$0.16	$0.02	$0.07	$0.02	$0.02	$0.02	$0.06	$0.37	$0.34
Mining (stoping and development).....	.99	.17	.34	.07	.09	.28	.38	2.32	2.12
Hoisting and transportation (underground).....	.47	.04	-	.04	-	-	.04	.59	.54
General underground expenses, including mine repairs.....	.52	.06	-	.04	-	.02	.11	.75	.68
Surface expense (directly applicable to under-ground operation).....	.37	.02	-	.03	-	-	-	.42	.38
Total.....	2.51	0.31	0.41	0.20	0.11	0.32	0.59	4.45	4.06

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TABLE 2. - Summary of costs in units of labor, power, and supplies, Gold Hill mine, Sept. 1, 1936 to Oct. 1, 1936

(Ore hoisted during period, dry tons, 3,573; ore and waste hoisted during period, dry tons, 3,919)

	Exploration	Mining and development	Total
Labor (man-hours per ton):			
Breaking, drilling, and blasting	0.13	0.80	0.93
Shoveling	.13	.83	.96
Haulage and hoisting	.03	.47	.50
Supervision	.04	.29	.33
General and repairs	.10	.58	.68
Total labor underground	0.43	2.97	3.40
Average tons per man per shift			2.35
Labor, percentage of total cost			63.00
Power and supplies:			
Explosives, pounds per ton of ore hoisted	.15	.77	.92
Timber lagging, feet board measure per ton of ore hoisted			9.27
Stulls linear feet per ton of ore hoisted			2.88
Total power,1/ horsepower-hours per ton of ore hoisted			50.05
Other supplies, percent of total supplies and power			36.00
Power and supplies, percent of total cost			37.00

1/ Total power purchased, including that used on surface and in the mill. Actual mine load equals 65 percent of the total power consumed.

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Grinding

Primary grinding is done in one 8-foot-diameter by 22-inch Hardinge ball mill, running at 18. r.p.m. The mill handles approximately 125 dry tons of new feed per day and is operated in closed circuit with a hydraulic trap and a Hummer V-32 electric vibrator screen, covered with M-378 Ton Cap steel screen cloth of approximately 35-mesh. The pulp is raised in a bucket elevator, 34 feet center to center, that travels 350 feet per minute.

The ball-mill discharge averages 65 percent of solids; no reagents or chemicals are added to the ore.

The average hourly capacity for September 1936 was 5.5 tons, and the ball mill operated 90 percent of the time. The daily average for the month was 119.3 tons.

Forged steel balls, 5 by 4 inches in size, are used. Ball consumption averages 2.25 pounds per ton of ore. Common, chilled, grey, cast-iron liners purchased from a local foundry are used. Liner consumption averages 0.63 pound per ton with a total cost of $0.028 per ton of ore treated. Liners are of the one-piece, bolted type.

The mill is direct-connected to a 150-horsepower induction motor through a spur gear and Link Belt silent chain drive.

Secondary grinding is done in a 4-foot-diameter by 3-foot Dewey ball mill, charged with 2-inch forged steel balls and running at 30 r.p.m. The feed to the small ball mill includes (See fig. 4) a concentrate from rougher Wilfley tables and the middling from two cleaner Wilfley tables. The concentrates are dewatered in a belt-drag classifier before being reground in the ball mill. Total feed to the regrind mill averages 1.25 tons per hour. Discharge pulp is kept at about 70 percent solids. Two-inch balls are added at the rate of approximately 40 pounds per day or 1.30 pounds per ton. Liners are made of local cast iron, and consumption is approximately 0.40 pound per ton. The mill is belt-driven by a 25-horsepower induction motor through a spur gear. A friction clutch takes care of starting overloads.

Gold Recovery

The original amalgamation practice was to pass all the minus 35-mesh ball-mill pulp through three hydraulic traps set in the main launder tandem fashion, over five rougher plates into five traps, and thence to five Wilfley tables. The reground concentrates were pumped back to another plate, through a trap, and thence to a Wilfley table.

After considerable experimentation it was found that the rougher plates were unnecessary, provided two plates and two tables were used in the regrind circuit. A better recovery also resulted by taking a high-grade concentrate cut from each regrind table and amalgamating it in drag mullers.

The chief reason for regrinding the gold in the small ball mill and drag mullers is to scour and polish the liberated gold particles.

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Amalgamation

The plates are cleaned each shift and dressed with ammonium chloride and clear water. The granulated ammonium chloride is sprinkled over the plates and vigorously brushed over the surface to loosen hard amalgam and free contaminating sulphides. After the plates have been cleaned and hosed the amalgam is softened with fresh quicksilver previously treated with metallic sodium. Excess amalgam is taken from the plates each shift. When the amalgam is soft enough so that it just slips under the pressure of the finger it is brushed lightly with whisk brooms dipped in a dilute solution of ammonium chloride. If the bed of amalgam on the plates is too thin after the soft amalgam has been scraped to the top of the plate with a section of rubber belting and removed with wall-scraping knives, then softened amalgam is added to the plate.

The dirty amalgam from the plates is ground in a hand muller with enough mercury to soften it. Each batch so treated is washed of all impurities and squeezed into pellets.

The high-grade cut from the regrind tables is ground out in drag mullers in the presence of mercury. The amalgam formed is removed from the mullers daily, cleaned, and pressed into small balls. The mercury loss in milling is 0.035 troy ounce per ton.

Hydraulic Traps

Eleven traps recover 40 percent of the gold saved. The traps, which operate by means of hydraulic pressure, are made on the job. (See fig. 5.) Each trap is cleaned daily, and the hutch product is ground with excess mercury in drag mullers. The amalgam is recovered and prepared as for the ground concentrates.

Concentration

Concentration is used in the present flow sheet to separate the more valuable sulphides from the gangue material. The coarse concentrate is subsequently classified and reground in the 4- by 3-foot ball mill to effect a greater separation of the gold, sulphide, and gangue particles.

As the four rougher plates are no longer used in the flow sheet, all of the screened mill pulp flows through the three traps in tandem set in the main launder, into the distributor, and thence through four traps in parallel to four standard Wilfley rougher tables. Each table is riffled with a modified-style no. 1623 riffle. Tables are actuated by 1-horsepower induction motors and run at speeds ranging from 265 to 275 strokes per minute.

Rougher tables make a rough concentrate which goes to the regrind circuit and a final tailing.

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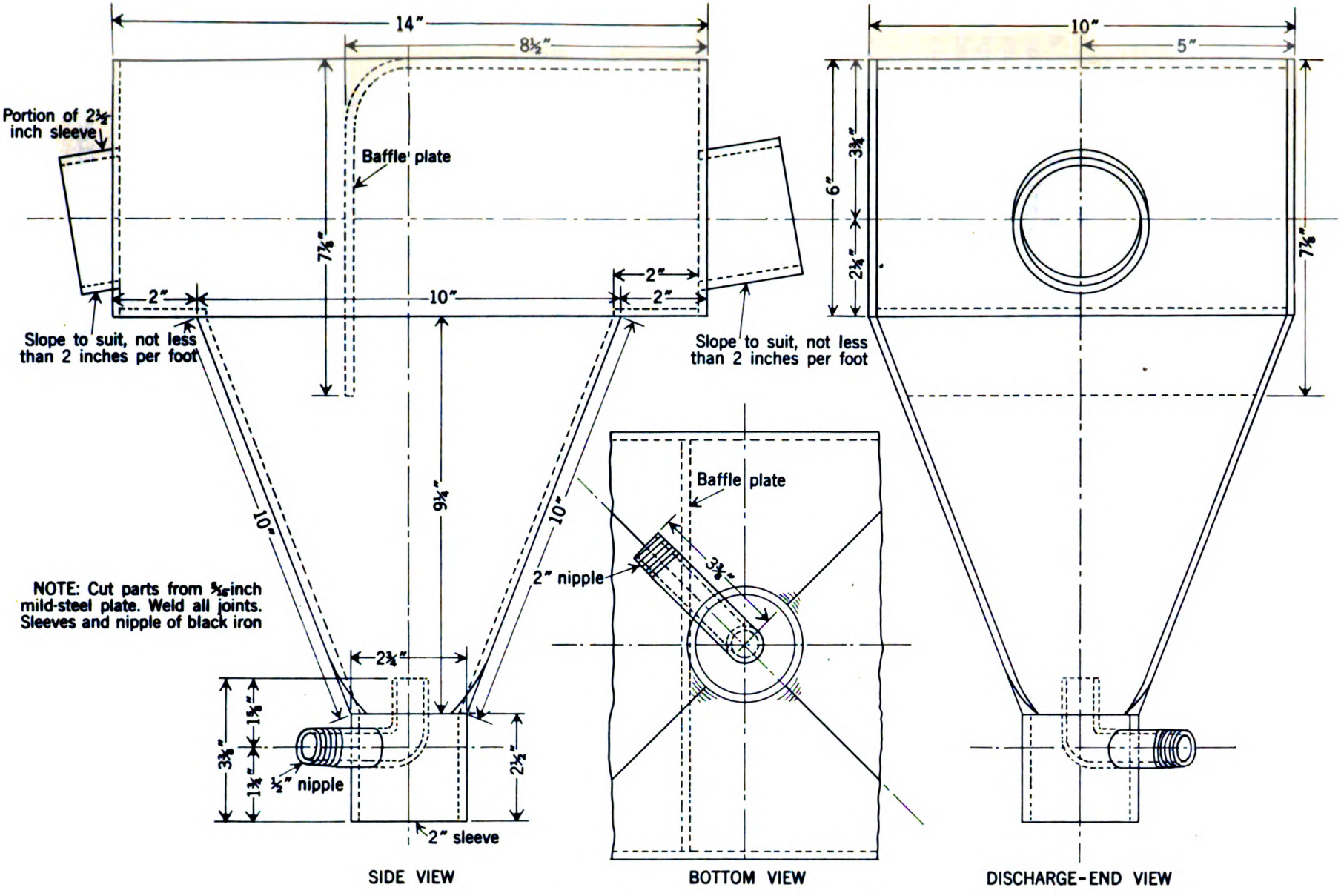

Figure 5.—Hydraulic gold trap, Talache Mines, Inc.

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Each regrind table makes three products: (1) A high-grade cut which is ground and amalgamated in drag mullers, (2) a rougher concentrate (middling) which is returned to the regrind circuit, and (3) a final tailing. One hydraulic trap is located in the circuit ahead of each table, and a large trap is located in the final tailing launder.

Retorting

When 400 to 600 ounces of amalgam has accumulated it is retorted in a no. 6 retort heated in an oil-fired furnace. The sponge is melted in a no. 16 graphite crucible with a flux of soda ash, borax, silica, and a small quantity of niter. The bullion is poured into a 3-1/4- by 3-1/4- by 6-1/2-inch cast-iron mold. Each bar is sampled by drilling two holes halfway through the bar on the center line, 1 inch from the end on opposite sides.

The bars range from 795 to 860 parts of gold per 1,000. The bullion is shipped by first-class mail to the mint. Postage and insurance total 3.8 cents per ounce, and mint charges are approximately 12 cents per ounce

Tailings Disposal

Tailings are carried by wooden launders and dumped directly into Granite Creek which flows past the mill site. No effort has been made to impound the tailings because of their small mineral content.

Sampling

Head samples are taken by hand every half hour and stored in a barrel. Each shift the sample is bucked down and sent to the assay-office dryer. Weight samples are taken by hand each half hour and combined at the end of the shift to calculate the tonnage handled. Tailing samples are also taken by hand every half hour.

Although hand sampling is awkward and inconvenient the results have checked closely actual mill results.

Labor

One mill foreman, 3 operators, and 3 crusher men comprise the mill crew. For special repair work or during the first of the month clean-up period, extra men are employed. The crusher men are employed two-thirds of the time in the mill and one-third of the time as waste-hoist operators.

The company operates a boarding house. Meals are $1.10 per day and rooms are $3.00 and $4.00 per month, depending on location. Married men live either in Placerville, a town 3 miles distant, or in Quartzburg.

Metallurgical and Operating Data

Metallurgical data are given in table 3 and operating data in table 4.

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TABLE 3. - Metallurgical data of the Talache mill, September 1936

Head assay: Gold.....ounces per ton.....	0.20
Tailing assay: Gold.....do.	.019
Total tons treated.....	3,579
Hours operated per day.....	24
Operating time, percent of possible total.....	90
Average tonnage per 24 hours.....	119.3
Gold recovery.....percent.....	90
Net water consumption.....gallons per ton ore.....	2,000
Amalgam recovered:	
Plates.....percent.....	8.6
High-grade cut from regrind tables..do.	40.1
Traps.....do.	41.5
Elevator and clean-up.....do.	9.8
Total.....	100.0

TABLE 4. - Operating data at the Talache mill for September 1936

Days operated.....		30
Men employed per 24 hours.....		6
Tons treated per man shift.....		19.8
Ball consumption, pounds of steel per ton of total ore treated:		
8-foot-diameter by 22-inch Hardinge mill.....	1.91	
4- by 3-foot Dewco.....	.34	2.25
Crusher operated.....hours.....		368

Cost of Milling

Table 5 gives direct and indirect costs per ton in detail; table 6 gives the costs of labor, supplies, power, and miscellaneous items for the various milling operations.

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TABLE 5. - Analysis of milling expense at the Talache mill for September 1936

	Cost per ton
Wages	$0.17
Grinding balls	.09
Liners	.03
Miscellaneous supplies	.065
Workmen's compensation insurance	.005
Taxes	.01
Building and equipment insurance	.015
Electric light	.01
Power	.125
Assaying	.02
General mine and mill administration	.01
Blacksmith shop	.01
Surface and general labor	.01
Mill repairs	.01
Mill administration	.05
Total	0.63

TABLE 6. - Operating costs at the Talache mill for September 1936

	Labor	Supplies	Power	Miscel-1/ laneous	Total
Crushing	$0.055	$0.03	$0.030	$0.015	$0.130
Fine grinding	.055	.15	.090	.050	.345
Amalgamation	.040	.01	.007	.005	.062
Concentration	.025	.01	.008	.005	.048
Superintendence	.050	.005	—	—	.055
Total	0.225	0.205	0.135	0.075	0.640

1/ Includes workmen's compensation insurance, taxes, building and equipment insurance, assaying expense, general mine and mill administration expense, and blacksmith-shop expense.

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